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PRECONDITIONING AND DESTRESSING SUMMARY OF FIELD INVESTIGATIONS

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PRECONDITIONING AND DESTRESSING: SUMMARY OF FIELD INVESTIGATIONS

by

Sylvain Dufresne*

SUMMARY

This report presents a literature survey of destressing and preconditioning practices used in underground mines throughout the world. From the thirty-one references studied, ten mines from Canada, the United States and South Africa were found to practice these techniques in highly stressed pillars, drifts or shafts.

Drilling and blasting parameters used at the mine sites are summarized along with the comments and recommendations made by the authors.

The review shows that preconditioning the rock mass can be successfully achieved. However, a high degree of risk is associated with the empirical approach currently used by mine operators and the results can often be completely opposite to the destressing objective sought. Further research is recommended for a more rational solution.

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Keywords: Rockburts; Preconditioning; Pillar Stability; Mining Methods.

RELAXATION DES CONTRAINTES INDUITES DANS LES MASSIFS ROCHEUX: SOMMAIRE DES ETUDES DE TERRAIN

par

Sylvain Dufresne*

SOMMAIRE

Ce rapport présente une revue de la littérature sur la pratique de la relaxation des contraintes par sautage utilisée dans des mines situées à travers le monde. Des trente-et-une références étudiées, dix mines canadiennes, américaines et sud-africaines ont fait appel à cette technique pour le minage de piliers, de galeries ou pour le fonçage de puits soumis à des contraintes élevées.

Le rapport résume les différents paramètres de forage et de sautage qui furent pratiqués. Les commentaires et les recommandations des différents auteurs sont également rapportés.

Cette revue permet de conclure que l'extraction en milieu soumis à des contraintes élevées peut être accomplie avec succès. Toutefois, la méthodologie utilisée par les opérateurs est empirique et présente des risques d'échec élevés. Des recherches additionnelles sont prescrites afin de proposer des approches plus rationnelles.

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Mots-clés: Coup de toit; Relaxation des contraintes induites; Stabilité des piliers de mine; Méthodes de minage.

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INTRODUCTION

Danger occuring from bumps or rockbursts is always a major problem in mines since it is unpredictable, creates production delays, influences productivity and threatens worker safety. Understanding of the triggering mechanisms has substantially progressed since the 1950's. However, practical demonstrations aimed at reducing the impact of the phenomena are mainly based on trial-and-error studies. Explosives have been successfully used to modify the stress field around excavations and safely re-direct stresses within the rock mass.

CANMET has already undertaken destressing research at its Elliot Lake Laboratory. Sigma Mines Ltd., Val D'or, Quebec, is now investigating the recovery of a highly stressed pillar. CANMET's scientists are cooperating by providing technical support and preparing field trials that will hopefully lead to a scientific methodology, instead of the present empirical one.

Part I of this report describes the main concepts underlying rock preconditioning. Under seven sub-headings, it describes the purpose of destressing, lists some design parameters, discusses computer modelling and describes some restrictions and risks associated with the practice. Finally, it reports observations and methodologies proposed by different authors whose work has been reviewed.

Part II describes several field experiments undertaken in Canada, the United States and South Africa. A comprehensive bibliography completes the report.

INTRODUCTION

Les dangers associés au minage dans des massifs rocheux susceptibles aux coups de toit demeurent un problème majeur pour les exploitations souterraines car ces évènements sont imprévisibles, ils induisent des délais dans la production, affectent la productivité et présentent un danger pour la sécurité des mineurs. Les mécanismes générateurs sont assez bien connus et sont discutés depuis les années 1950. Toutefois, les démonstrations pratiques sont principalement basées sur des méthodes par essais et erreurs. La relaxation des contraintes à l'aide d'explosifs est alors pratiquée et elle tente de diriger les contraintes concentrées près des excavations vers des zones ne présentant aucun danger.

CANMET a contribué aux recherches dans ce domaine par l'intermédiaire de son laboratoire d'Elliot Lake. Les Mines Sigma de Val D'Or, Québec, étudient présentement l'exploitation d'un chantier soumis à des contraintes élevées. Les chercheurs de CANMET sont particulièrement intéressés de fournir une expertise technique tout au long du projet, mais également à préparer des essais de terrains qui, il est espéré, pourront servir à l'élaboration d'une approche scientifique au problème qui remplacerait l'approche empirique actuelle.

La première partie du rapport présente les principes généraux de la relaxation des contraintes. Sept chapitres traitent entre autre des paramètres de conception, discutent de la modélisation numérique et décrivent certains risques et restrictions liés à sa pratique. Des commentaires et de nouvelles méthodologies d'analyse rapportés par les différents auteurs sont décrits.

La seconde partie décrit les essais de terrain réalisés dans des mines du Canada, des Etats-Unis et de l'Afrique du Sud. Une bibliographie complète ce rapport.

PART I

CONCEPTS

1. Purpose of Destressing

One definition of rockbursting is the sudden failure of rock characterized by the breaking up and expulsion of rock from its surroundings, accompanied by the violent release of energy. This phenomenon happens when a volume of rock is strained beyond its elastic limit. The failure is of such a nature that accumulated energy is released instantaneously.

Destressing or preconditioning the rock mass aims to release this excess energy in a controlled manner by reducing the stiffness of the stressed region and allowing closure and displacement of the rock structures to occur. Drilling and blasting techniques are usually used for this purpose. Destressing has been applied on a regular basis for the last 20 to 25 years, with some apparent success.

A detailed scientific explanation of how destressing results in a decrease in pillar stiffness is not yet available. Since there is no free face, we must assume that the creation of new fracture surfaces is not important, and that separation or dislocation of fracture surfaces is more likely to occur (Board et al., 1983).

The following sections discuss the main conclusions proposed by different authors on various topics related to destressing.

2. Design Parameters

A variety of design parameters influence the success or failure of preconditioning.

Hole spacing can be determined by powder factor calculation, peak particle velocity analysis or experimental investigation. All these methods are derived from observations of fracturation under a given set of conditions. If the in situ environment changes, spacing selection may no longer be valid.

It was observed from blast investigations that no significant differences in damage to the rock occurred when different powder factors were used. It is therefore not appropriate to use the powder factor calculation for destress blast design since it only considers the weight of explosive (Cullen, 1988).

It was also noticed that larger holes give faster detonation velocities, greater detonation and explosive pressures. However, there is a limit beyond which increasing detonation energy results in no increase in the fracture extension. In situ studies provide the most reliable assessment to decide which explosive is best suited for destress blasting (Cullen, 1988). Good stemming is essential, but it's length depends on the in situ results.

From a geomechanical point of view, the main parameters governing preconditioning are discontinuity orientation, existing in situ stresses and the degree of confinement (Cullen, 1988). Structure investigation (shear planes) is of particular interest since it helps to understand the geological forces involved and it helps in the selection of the hole length that will be required to destress the critical structures (Roux et al., 1957).

3. Computer Modelling

Computer modelling by Karworski et al., (1979) suggests that in-vein blasting is preferred; multiple initiation points ensure complete detonation and eliminate the possibility of mining into residual explosives. Also, the best results are obtained when detonation occurs simultaneously with the passage of the detonation pressure wave from the adjacent hole; this delay timing can be achieved with pressure sensitive detonators. However, safety considerations limit their use; high milli-second delay detonators or conventional detonators with equal delay can also be used (Cullen, 1988).

Crouch (1972) studied a hypothetical approach and concludes that destressing should be performed when the vein material is near the point of failure. If it is performed too early, the material will temporarily unload and will still be susceptible to burst.

A 3-D finite-element analysis (MULSIM) was used in a coal mine to evaluate the stress redistribution patterns resulting from destressing a longwall face (Haramy et al., 1988).

4. Restrictions to the Method

Some limitations to preconditioning are caused by hand-held drilling equipment which restricts the size and length of holes. It then becomes normal practice to wait until the pillar is small before destressing, resulting in increased exposure time to possible on-shift bursting. Other limitations concerning subsequent mining are related to the hazardous conditions of misfires occurring during destressing (in the vein). Timing is a critical parameter (bursting and exposure time). Destressing involves

production delays and loading must be performed properly (Blake, 1984).

5. Risk

Preconditioning involves some risk and high costs. Careful planning and very close monitoring are required. The technique is labor and equipment intensive (McMahon, 1988). Preconditioning effort can be quite successful but it may result in creating another and perhaps even worse stress problem. Efforts must be carefully studied and engineered (Blake, 1982).

Dangerous high-stress conditions may occur in coal mining if portions of the longwall face are destressed while isolated areas are left untreated (Haramy et al., 1988).

6. Reported Observations

Blake (1982) concluded that preconditioning the vein was more effective than preconditioning the wall rock. This paper provides some specifications of the drilling and blasting equipment used at the mine site. He also noticed less seismic events when mining was performed in a preconditioned rock. A visual study of the holes after blasting revealed that 102 mm diameter holes were now some 200 to 300 mm in diameter, surrounded by a crushed zone extending out another 150 mm. No pattern of radial cracks around the crushed zone was observed but the entire vein appeared highly fractured.

Although the instrumentation program provided useful information, it appeared to be of little use in evaluating the effectiveness of preconditioning (Blake, 1982).

Blake (1984) reports an experiment which shows that the stronger the destress blast, the more effective it is. With 38 mm diameter holes, spacing should not exceed 1.6 m; with 100 mm diameter holes, spacing can be in the 3.2 m range. The best practice is the trial-and-error method.

Following a destress blast test, Roux et al., (1957) observed that prononced fractures sometimes ran to a distance of 0.46 m.

Cullen and Scoble (1989) identified from field measurements, relationships between fracture frequency and elastic modulus, blasthole spacing and fracture frequency, and between blasthole spacing and elastic modulus.

Diamond drilling can be used to evaluate high stress zones. Core disking takes place when the drill bit approaches such a zone,

and pieces of grit are found in the core when the main high stress zone is crossed (Roux et al., 1957).

7. Proposed Methodologies

Cullen and Scoble (1989) proposed a methodology for destress design. They suggest that destressing should be performed on two rows, the second one located to create a 30 degree weakness plane to the maximum principal stress.

Destressing in a shaft is reported by Jeremic (1982). The author suggests, as a rule of thumb, that a destress blast should more or less double the number of joints and cracks that were initially present in the rock. Damage to the rock resulting from the influence of destress holes was in the range of 0.8 - 3.9 m.

Willan et al., (1985) lists tables presenting data coming from different mine site investigations (drilling and blasting patterns,...). This paper also suggests a destressing methodology illustrated by Figure 1.

Haramy et al., (1988) report studies done by the USBM where expansion grout is used to induce rock fracturing. The mix of lime compounds and water is confined in a small borehole and an expansion force of up to 62 MPa, generated by a hydration reaction, was recorded (increasing rate of 13.8 MPa/day).

PART II

FIELD EXPERIMENTS

1. Cambell Red Lake Mine

1.1 - 1102E 'B' Crown Pillar (Makuch et al., 1987)

A cut-and-fill method is used on the 10th level (approximately 450 m depth). The pillar length and height are 77 m and 8.2 m respectively. (Refer to Tables 1 to 3 for a summary of the design parameters from the mines under study - Sections 8 to 15). The average stope width is 2 m.

Stoper and extension steel are used to drill holes from the stope at an average spacing of 1.8 m within 1.5 m of rail elevation. Zero delays were used except for the portion designed to break, where 25 ms delays were used. ANFO with a powder factor of 0.20 Kg/cu m was used for blasting.

Closure and excessive damage to drift floor and track were measured. Stress transfer or blast shock waves lead to boxhole pillar burst.

1.2 - 1421 W Boxhole Pillar (Makuch et al., 1987)

Shrinkage mining was praticed. Chutes are located at every 8 m. Seven boxholes distributed over a length of 48 m; three 45 mm holes were drilled in each of them.

ANFO blasting agent with a powder factor of 0.09 Kg/cu m was selected. A 25 ms/hole delay initiated the blast.

Little added stress to the surrounding boxhole was measured and it was assumed that load was transferred to the large pillar below the level. Closure was measured at all stations and a subsequent burst proved that at least one boxhole remained intact.

1.3 - 1902 E Crown - 1802 Sill Pillar (Makuch et al., 1987)

Cut-and-fill was practiced until the remaining pillar was 6 m high. The 1802 pillar was 31 m long and the 1902 pillar was 44 m long.

Six meter long, 44 mm diameter upholes were drilled in the stope. They were drilled 1.8 m apart. The pillar was 7.5 m high and the powder factor was 0.20 Kg/cu m. ANFO and delays of 25 ms were used in each hole. The top 1.5 m of the holes was not loaded.

Small events were measured for a period of 30 minutes after the blast. A rockburst occurred in the upper part of the pillar and microseismic activities were recorded for 4 hours. Significant damage was surveyed on the 18th level back. Nevertheless, walls were in fairly good condition and another burst was detected after the production blast.

1.4 - 1604 E Crown Pillar (Makuch et al., 1987)

A vein 1.8 m wide with a strike of 150-180 m was mined with 10 m high stopes.

Holes were drilled in a vein parallel to the main zone, creating a shadow pillar effect. A longhole machine was used to drill 45 mm holes, 1.4 m apart. A powder factor of 0.7 Kg/cu m (normal production blast: 2.1 Kg/cu m) was planned. ANFO was used in holes up to 1.5 m of the collar. Zero delay Nonels were selected.

Closures were roughly estimated by computer simulation. Seismic activity was measured in the abutment near the pillar. The instrumentation located near the blasting site was in good condition after the blast. The blast lowered the elastic modulus by 25%-50%. No ground problems were recorded during recovery of the pillar.

1.5 - Headings (CANMET's Elliot Lake Lab.)

The Elliot Lake Laboratory of the Mining Research Laboratories proposed a procedure for destressing development headings and stope sills. These can be summarized as follows:

Face: 2 holes half way up on face and 1/3 width in from walls;

Flat holes: load 1 of 3 m holes for 1.8 m advance
1.2 of 3.7 m holes for 2.4 m advance
1.5 m of 4.3 - 5.5 m holes for 3 - 4.3 m advance.

Wall: 4 holes, one in each corner of face. Drill up and down 30 degrees, angle out 45 degrees.

From advance direction;
Load 1 m of 2.4 m holes for 1.8 m drive advance.
1.2 m of 3 m holes for 2.4 m advance.
1.5 of 3.7 m holes for 3 to 3.7 m advance.

Destressing holes must be fired before the first round shot.

1.6 - 1470 Drift (Scoble et al., 1987)

The test procedure is as follow: 45 mm diameter horizontal holes were drilled normal to the drift to a depth of 2.5 m. Drillhole spacing was successively increased from 0.3 m to 0.6 m, 0.9 m and 1.2 m. The 1.2 m bottom part of the holes were loaded with AMEX (density of 0.98 g/cc). The #0, #2, #4, #6 and #8 Nonels are used.

Test holes were drilled between each blasted hole, before and after the blast (NQ - 76 mm).

A relationship between fracture frequency and distance between boreholes was established. A reduction of elastic modulus with fracture frequency increase was noticed. Blast fracturing detected by impression packer and core logging revealed that rock damage extended up to 16 diameters (1.4 m); dilatometer readings produced values as high as 1.6 m.

It was also observed that the fractures were mostly oriented in the pre-existing discontinuity direction. With a 1.5 m hole spacing, no fracturing should be expected at midway between holes. A relationship between radial distance from blast, weight of explosive and peak particle velocity is proposed.

1.7 - 1704 EW Stope (Cullen, 1988)

A study of this stope was undertaken by investigating 10 breast blasts, 4 without destress holes, 3 with one (1) back destress hole (0.6-0.8 m from roof) and 3 with one (1) center destress hole.

The face dimensions were: width between 1.5-2.4 m, height between 2.4-2.8 m and length of 3 m. The destress holes were 4.5 m long. Destress hole diameters were 32 mm. ANFO and safety fuses were used.

The survey instrumentation included a microseismic monitoring system, tape extensometer, visual inspection, vibrating wire gauges, fill cells and stope convergence meters.

It was noticed, after the blast, that miners had difficulty drilling below the destress holes. Damage to the stope back due to destress holes occurred in only one case. Minimum back damage was recorded with a center destress hole.

A circular crushed zone up to 30 mm around blast holes was identified and apparently not affected by pre-existing discontinuities or the stress regime. The fracture zone was detected up to 100 mm around the blast holes, aligned along pre-existing discontinuity planes. Influence of in situ stresses was not determined. Oval holes with the long axis parallel to the longest fractures were identified.

It is concluded that a 4.5 m destress hole in a 3 m production round was not an effective means of destressing. The comments presented in this investigation should be regarded as site specific observations.

1.8 - 1670 W Stope Crown Pillar (Cullen , 1988)

A natural fractured zone was identified around a 10 m thick pillar. The zone was 2 m deep at the top of the pillar and 2.5 m thick at the bottom.

2. Macassa Mine (Hanson et al., 1987)

2.1 - 58-40 Crown Pillar

Fourteen holes were drilled by bar and arm set-up. The holes were 63.5 mm in diameter, with 3 m spacing, 16.8 m in length and located in the footwall. ANFO was used as the explosive with barite stemming or inert gel on a length of 3.6 m from the collar being used to confine the blast energy.

A powder factor of about 0.14 Kg/cu.m. (assuming a density of 2.7 t/cu.m.) was selected. Holes were initiated with millisecond delays (42.2 Kg/delay).

Computer modelling was used to predict the yielding of ground, closures and peak particle velocities.

Blasting occurred during a long weekend. Two misfires, one at each end, resulted in a powder factor of 0.03 Kg/tonne. Some cratering due to poor stemming was noticed.

Diamond drilling through the central core of the pillar revealed that the central part of it was highly fractured in the plane of the destress holes. No more disking was present on either side of this zone.

Convergence of up to 28 mm was measured. Destressing this pillar was considered to be a success. However, assessing the correct moment to destress is not always a simple matter.

None of the individual investigation methods were foolproof in preventing bursting occurrences (U/G convergence measurements, microseismic source location and computer simulation).

A much higher powder factor should be used for full pillar failure.

3. Falconbridge's Sudbury Operation (Slade, 1968)

Three meter destressing holes were drilled in the hanging wall and blasted at each breast.

Destressing in a cut-and-fill 18.3 m high pillar was done with thirteen 18.3 m diamond drill holes, 6.1 m apart, down into the hanging wall. The bottom of the holes were 6.1 m away from the vein. Six meters of high explosives was placed in each hole and blasted simultaneously. Results were apparently successful.

4. Thierry Mine (Jeremic, 1982)

The best results for this shaft sinking project were obtained when the entire blast was confined. Two separate charges were used as counter blast (one charge close to the collar acted as stemming). Both were of equal power.

The pattern was composed of four 6 m depth vertical destressing holes located at 0.6 m from the corners.

5. Coeur d'Alene Mining District (Blake, 1972, 1982, 1984, CSM-Blake, 1972, Board et al., 1983, McMahon, 1988)

5.1 - Galena Mine, Idaho

Monitoring of the pillar started when it was 18.3 m thick. Critical bursting started when it approached 10.7 m thick and increasing seismic velocity was noticed when the pillar was reduced from 18.3 m to 6.1 m.

Broad-band microseismic monitoring and seismic velocity surveys were used and the mechanics of destressing were determined by numerical modelling.

A trial and error approach was used. The best hole configuration is when the spacing is uniform. For destressing to be effective, it must be done at the right place, at the right time (!).

Six meter, 48 mm diameter long holes were drilled on 1.5 m centers, loaded with ANFO prill to within 1.5 m of the sill and 1 m of the collar, and were fired with millisecond delays. Figure 2 shows the drilling pattern of the destress holes.

Blasting was performed on a single line of long holes in the footwall of the vein. Destressing was successfully performed. The seismic velocity was significantly reduced when the pillar reached a thickness of 3 meters. Figure 3 shows the velocity surveys (before and after destressing).

5.2 - Hecla's Star Mine: 1st demonstration

The test site is located 2350 m below surface and is 61 m long. The vein was preconditioned for 12 m above and below the level. An initial test pattern was drilled to verify the 3 m burden between the hole bottoms. A few test blasts established that a hole separation of 2.3 m would produce the best results.

Destressing crosscut #7 was performed 6.1 m away from the vein, in the quartzite wall. A test hole of 55 Kg of high explosives revealed that each round would require a shift or more of cleanup and there was a possibility of knocking the raise timbering. It was then decided to drill the destress holes directly in the vein for crosscut #10. Figures 4 and 5 show a plan view and the blasting characteristics respectively.

Thirty metre long holes, 102 mm in diameter were drilled. Some 1372 m of drilling (blasting and instrumentation holes) and 4082 Kg of high explosives were used in both #7 and #10 crosscuts. The bit diameters were 92 mm and 102 mm. Drilling in #10 crosscut (destressing in the vein) was done without ground control problems (destress holes used for the burns in following production cuts).

The blasting procedures and the choice of the blasting agents were essentially based on field trials since there is no published or reported guidelines. Previous destress blasting experiences practiced in the Coeur d'Alene mines showed that the faster and more powerful the explosive, the better the results in terms of destressing. A high-velocity water gel was selected as the primary explosive. It appeared that water gel produced the sharpest detonation and poured water gel produced the weakest one.

The maximum density of explosives was 7 Kg/m (explosive density of 0.95 g/cc) and the holes where stemmed with water or a clay-sand mix, and detonated with 100 ms delays.

The 6.1 m of unloaded hole length resulted in little cratering of the crosscut. Detonating cord was used along the entire length of all holes (for safety precaution) but it did reduce the effectiveness of tamped clay-sand stemming in the upholes. A typical view of loaded holes is shown in Figure 6.

Seismic velocity, stress, fill pressure and displacement monitoring were performed in the #7 and #10 crosscuts. They confirmed that destressing was successful. Also, finite-element computer simulations of blasting were done.

A reduction in the horizontal component of stress was observed. Some popping rock and typical difficult conditions occurred when mining resumed in the #7 crosscut (destressing in the wall). The wall was badly cracked, creating ground control and dilution problems. A rockburst in the #7 crosscut occurred after the fourth cut.

Much lower seismic activity was measured in the #10 crosscut. Seismic activity and rockbursting resumed when mining was performed above the preconditioned zone.

5.3 - Hecla's Star Mine: 2nd demonstration

This investigation comprises 8 steps (Figure 7): 1: Mine 7700 level stopes (Section 12.2); 2: Phase 1 preconditioning in 7900 #8 crosscut; 3: Phase 1 preconditioning in 7900 #12 crosscut; 4: Mine #8 and #12 raiseups on 7900 level; 5: Destress drilling and blasting from #9 crosscut on 7500 level; 6: Destress drilling and blasting from #5 crosscut on 7500 level; 7: Complete second floor mining in #8 and #12 stopes on 7900 level; 8: Phase 2 preconditioning from second floor in #8 and #12 stopes on the 7900 level.

The 7500 level pillar had already experienced a rockburst (magnitude of 2.6 on the Richter scale). Figure 8 shows the drilling pattern and Figure 9 shows the blasting characteristics of the preconditioning of 15 meters of the rock mass above and below crosscuts #8 and #12.

Phase 1 included the preconditioning of 140 m in strike length of the vein (7900 level). Some 2212 m of holes were drilled and 4570 Kg of Tovex 5000 where used. Phase 2 included the drilling of 910 m of holes (7500 level) that were blasted with 1820 Kg of explosives (Figure 10: destress holes from #9 crosscut).

After destressing, mining resumed with moderate microseismic activity and no burst occurred until the #8 stope top was reached, where a rockburst of 1.9 (Richter magnitude scale) was measured.

The measurements revealed that few microseismic events accompanied the mining of the preconditioned zone, compared to the 20 bursts and high microseismic activity that occurred while mining above the preconditioned zone of the 7700 level (Section 12.2). Large-scale stope preconditioning was found to be an effective means of rockburst control.

5.4 - Hecla's Star Mine: Grouse Vein

A pillar between 2040 m and 2100 m below surface was mined by the underhand cut-and-fill method.

Holes 29, 50 and 57 mm in diameter, 5 to 8 m in length, were drilled with jackleg drills on 2 to 3 m center upward patterns, into the above pillar. To prevent mining into fractured ground, no hole was drilled directly at future raise locations (Figure 11).

The holes were loaded with ANFO or watergel, depending on the hole's condition; in either case, a cast primer was first placed in the hole, with 200 grain detonation cord. Explosives were loaded to within 1 m of the collar. Non-electric millisecond delay primers were used. The poor loading of the holes resulted in the detonation of only 180 Kg of explosives, or about 1.5 Kg/m of hole.

Results of destress blasts were immediately evident from the instrumentations and observations. Seismic activity was minimal, closure was evident, an increase in the cemented fill horizontal pressure was recorded, as was a decrease in the pillar stress. Minor sloughing resulted from a minor burst in the stope wall.

Large amounts of wall closure indicated the success of the test (102 mm in first cut, and 350 mm more in the raise area during the following year). When the pillar height was reduced to 10 m, closure rate was recorded to be extremely high and instrumentation showed a decrease in the stresses.

After destressing, some problems made raise driving difficult, due to loose rock and overloaded blasting rounds. Even if bad backs were encountered, raises were driven without bursts. This operation was normally very hazardous.

5.5 - Coeur d'Alene Terra Tek Mine (Jenkins et al., 1983)

The mining method was converted from overhand cut-and-fill, to underhand cut-and-fill coupled with the preconditioning of the remaining pillar.

The original preconditioning design was: 10.7 m long, 57 mm diameter vertical boreholes, drilled upward and at every 3 m, the entire length of the stope block (drilled with jacklegs, 51 mm and 64 mm cross bits). A total of 29 holes were drilled.

Driving raises (for underhand cut-and-fill) was difficult for 2 of the 3 raises (one resulting from inexperienced miners overloading their raise rounds). High closure rates required extensive raise timber repairs. This attempt was considered to be a success.

6. East Rand Proprietary Mine

6.1 - Practical Studies (Part I: Roux et al., 1957; Part II: Hill et al., 1957)

The practice is to drill 3 m deep destress holes, 1.5 m apart in reef faces and 1 m apart in dyke faces. When assuming normal working face advance at the rate of 1 to 1.2 meter per week, this procedure ensures that a cushion of rock at least 1 to 1.5 m thick always exists at the face.

Fifty percent of each hole was charged with 60% ammon-gelignite (70% in dykes), or with 60% Dynagel, the rest of the hole was completed with sand tamping.

Some studies examined the influence of destressing on the incidence and severity of bursts, the delays and the effects on production, on hanging wall conditions, and the time of the day at which bursting occurred.

From the cases studied, these authors concluded that the destressing practice gave good results but the technique needs to be studied in more detail. However, it is not effective where dykes or faults were present and an alternative approach is required to prevent bursts when such structures exist.

If the decision is taken to destress a reef face, then the entire face will be destressed, since destressing only a portion of it tends to concentrate the stress on the under-stressed portion, and thus to increase the danger of bursting.

6.2 - Comments on Roux et al paper (Part I and II) (Roux et al., 1957)

In hard and tough dykes which could not be fractured with 50% of each hole charged with 60% Dynagel (Section 6.1), 3 m long holes were drilled in sets of three, but only 200 mm apart. The two outer holes were charged for 70% of their length. The method was successful in a 6 m wide residual strip of dyke but not in a 21 m wide dyke in a stope. The holes were detonated sequentially. New trials where all holes were detonated simultaneously appeared to be more successful.

Destressing of a shaft pillar was also performed. Three meter long destress holes were drilled, straight at the face, midway between the walls, 1.5 m apart, along the length of the face. The back 0.6 m of the holes were charged with 40% gelignite and the holes were tamped normally. This blast was a failure; re-charging with 60% gelignite and fully tamped holes revealed to be a success. Hole length was then reduced to 2.4 m, but the design subsequently returned to 3 m since it was less expensive. Explosive charge of the holes was reduced to 50% of the total length; this was unsuccessful. The holes were gradually moved closer to the footwall to prevent damaging the hangingwall.

The destressing practice is used every time it is needed. It is also performed in the development headings.

7. Creighton Mine No.11 Shaft (Garrood, 1982)

This approach to control rockbursts has been largely empirical. Destressing was practiced in drifts with the mining advance.

Two 7.3 m long destress holes (double length of the round) were drilled in the drift heading and two 3.7 m long holes were drilled at an angle of 40 degrees from the advance direction, in each shoulder.

The full length of the 7.3 m holes and the bottom 1.5 m of the 3.7 m holes were blasted with the rest of the round. Sometimes, three additional 3.7 m holes are drilled in the back, at an angle of 40 degrees; these were collared at the top of the face. The bottom 1.2 m of each hole is also blasted with the round.

8. Coal Mines and Destressing (Haramy et al., 1988, and USBM RI 9225)

Destressing is accomplished using 51 mm holes, 4 to 4.5 m long, at approximately 1.2 m centers. Their depth was dependent on the daily advance and on the location and magnitude of the stress abutment (postulated) ahead of the face.

An auger drill was used in highly stressed areas to weaken the seam and cause failure of the coal at a reduced stress level (152 mm diameter holes, maintained not less than 10 m long ahead of the face). The use of barricades between drill and rock is essential because some bursting occurs while drilling (this method is also applied in German mines - Will, 1984).

Hydraulic fracturing is used in porous mediums (test reported to be successful with a fluid pressure up to 28 MPa). However, this technique is time-consuming.

CONCLUSION

A literature survey on destressing and preconditioning has been undertaken. The main concepts underlying the technique and the main conclusions of the investigators are reported. The field investigations are also briefly described.

The general perception is that preconditioning of the rock mass can be successfully achieved. However, the technique is empirical and site specific, and presents a high degree of risk. Development of a more rational solution must therefore be undertaken.

Several authors emphasized the difficulty of identifying the parameters controlling rockburst occurrences and the lack of investigative "tools". Preconditioning the rock mass is perceived as a solution to the problem which is, however, not yet fully understood. Drilling and blasting layouts are based on empirical practices and large scale pillar destressing is, in a certain respect, a first/last attempt.

To address this particular problem and propose to the mine operators a solution for the recovery of highly stressed pillars, research must be undertaken to elaborate a scientific approach based on technical information provided by the greater use of data gathering instrumentation. Such approach will improve, among other things, drillhole diameter selection, hole location and orientation, and also allowing to plan a better explosive detonation sequence.

This research project would involve field investigations where full instrumentation of the rock mass is carried out. The rock mass behaviour study, executed before, during and after the preconditioning process would be linked to numerical modelling. Repeating the study for several scenarios would lead to the scientific evaluation of the influence of the main drilling and blasting parameters. A process of this type is suggested for future studies.

ACKNOWLEDGEMENTS

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Table 1: Principal design parameters from the mines under study.

Section	Mine	Pillar					Mining method
		Name	Depth (m)	H (m)	L (m)	W (m)	
8.1	CRLM	1102E'B'	450	8,2	77	2	Cut & fill
8.2	CRLM	1421W			48		Shrinkage/Boxholes (7) every 8 m
8.3	CRLM	1902-1802		7,5	44/31		
8.4	CRLM	1604E			150-180	1,8	10 m high stopes
8.6	CRLM	1470					
9.1	Macassa	58-40					
10	Falconbrg			18,3			Cut & fill
12.1	Galena						
12.2	Star	#7,#10	2350	80	61		Cut & fill
12.3	Star	#8,9&12	2400	60	140		Cut & fill
12.4	Star	Grouse	2050				Cut & fill
12.5	Terra Tek						Undercut & fill

Table 2: Principal design parameters from the mines under study.

Mine	Pillar name	Holes						Explosives	
		Location	No	Length (m)	Dia. (mm)	Spacing (m)	Orient.	Type	PF (Kg/cu.m)
CRLM	1102E'B'				45	1,8		ANFO	0,2
CRLM	1421W	Chutes			45			ANFO	0,09
CRLM	1902-1802	Stope		6	44	1,8	Up	ANFO	0,2
CRLM	1604E	Vein			45	1,4		ANFO	0,7
CRLM	1470	Normal to drift		2,5	45	0,3 to 1,2	flat	AMEX	
Macassa	58-40	Footwall	14	16,8	63,5	3		ANFO	0,14
Falconbrg		Hanging wall	13	18,3		6,1	down		
Galena		Footwall		6	48	1,5		ANFO	
Star	#7	6,1m off vein		30	102	2,3			
Star	#10	Vein		30	102	2,3			
Star	#8,9&12							Tovex 5000	
Star	Grouse			5 to 8	29;50; 57	2 to 3	up	ANFO, gel	
Terra Tek			29	10,7	57	3	up		

Table 3: Principal design parameters from the mines under study.

Mine	Pillar name	Stemming length (m)	Delays (msec)	Comments
CRLM	1102E'B'		0, 25	
CRLM	1421W		25	
CRLM	1902-1802	1,5	25	
CRLM	1604E	1,5	0	Holes parallel to main zone: shadow effect. Normal production PF = 2,1 Kg/cu.m.
CRLM	1470	1,3	0, 2, 4, 6, 8	
Macassa	58-40	3,6		Misfires: PF=0,03 Kg/cu.m.
Falconbrg		12,3		Diamond drill holes for destressing, blasted simultaneously.
Galena		1		Figure 2. Holes performed on a single line.
Star	#7,#10	6,1	100	Figures 4, 5 & 6. Total 1372 m of drilling and 4082 Kg of explosives. Maximum explosive density: 7 Kg/m.
Star	#8,9&12			Figures 8 & 9. Total 3122 m of drilling and 6390 Kg of explosives.
Star	Grouse	1		Figure 11. No hole in future raise locations. 180 Kg of explosives (1,5 Kg/m).
Terra Tek				

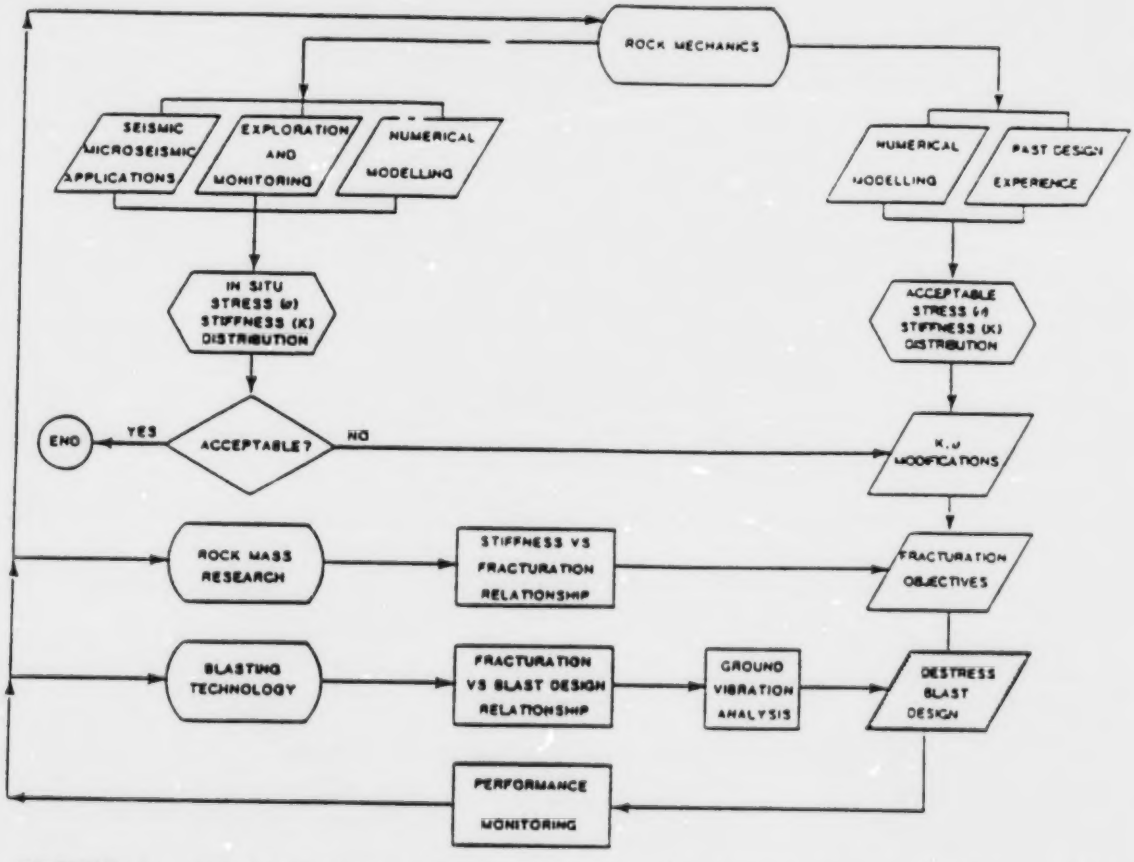

FIGURE 1: Engineering approach to destress blast design (after Willan, 1985).

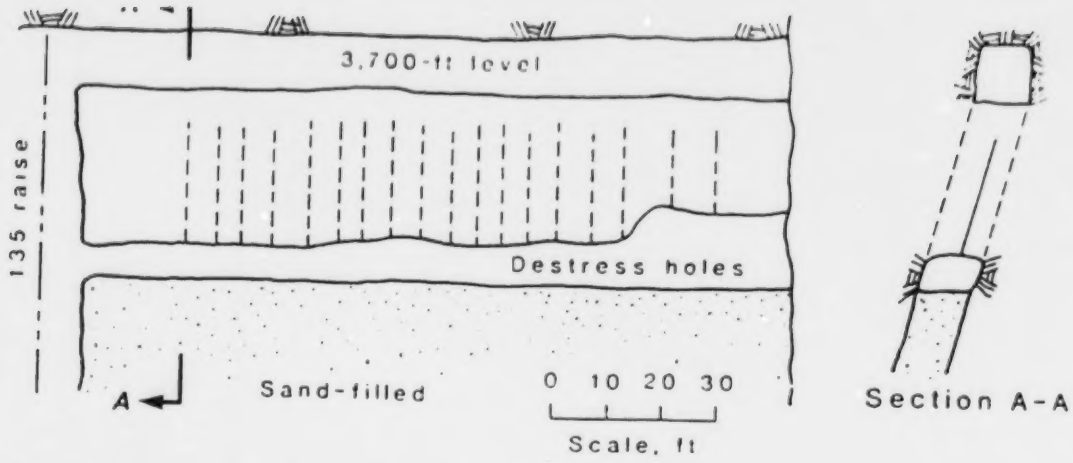

FIGURE 2: Holes drilled into pillar (after McMahon, 1988).

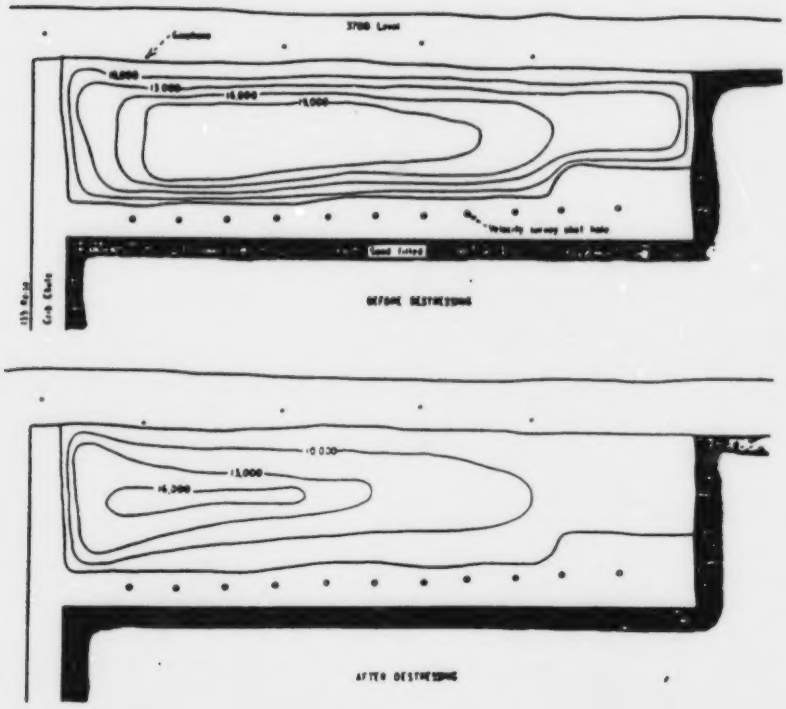

FIGURE 3: Seismic velocity contour plots (after Blake, 1972).

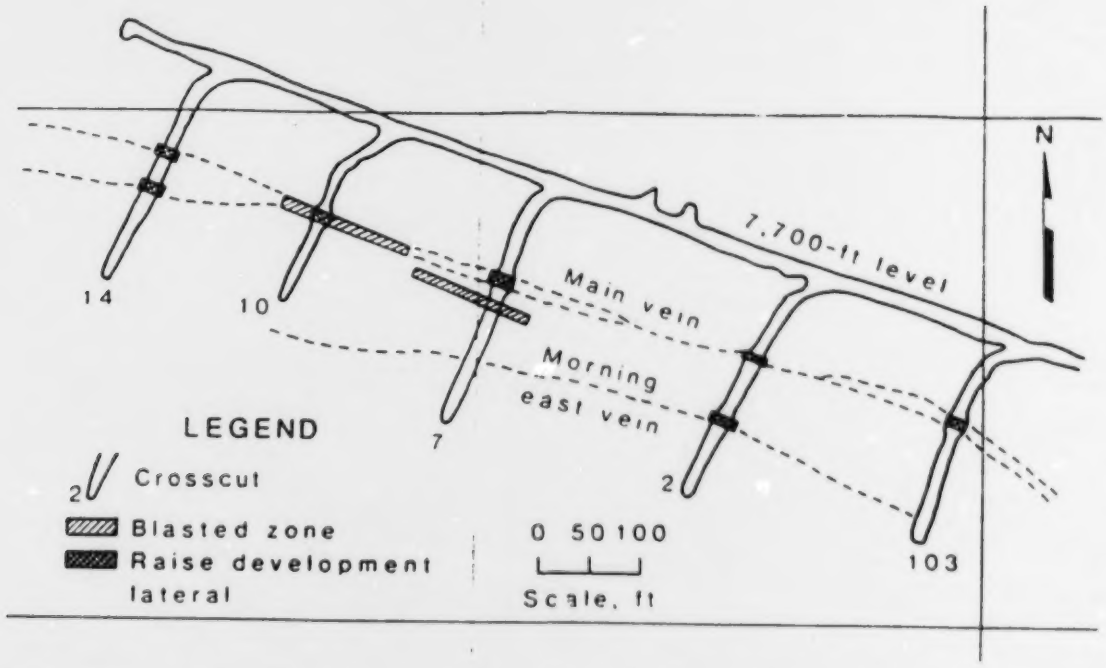

FIGURE 4: Plan view of preconditioned zones of the 7700 level (after McMahon, 1988).

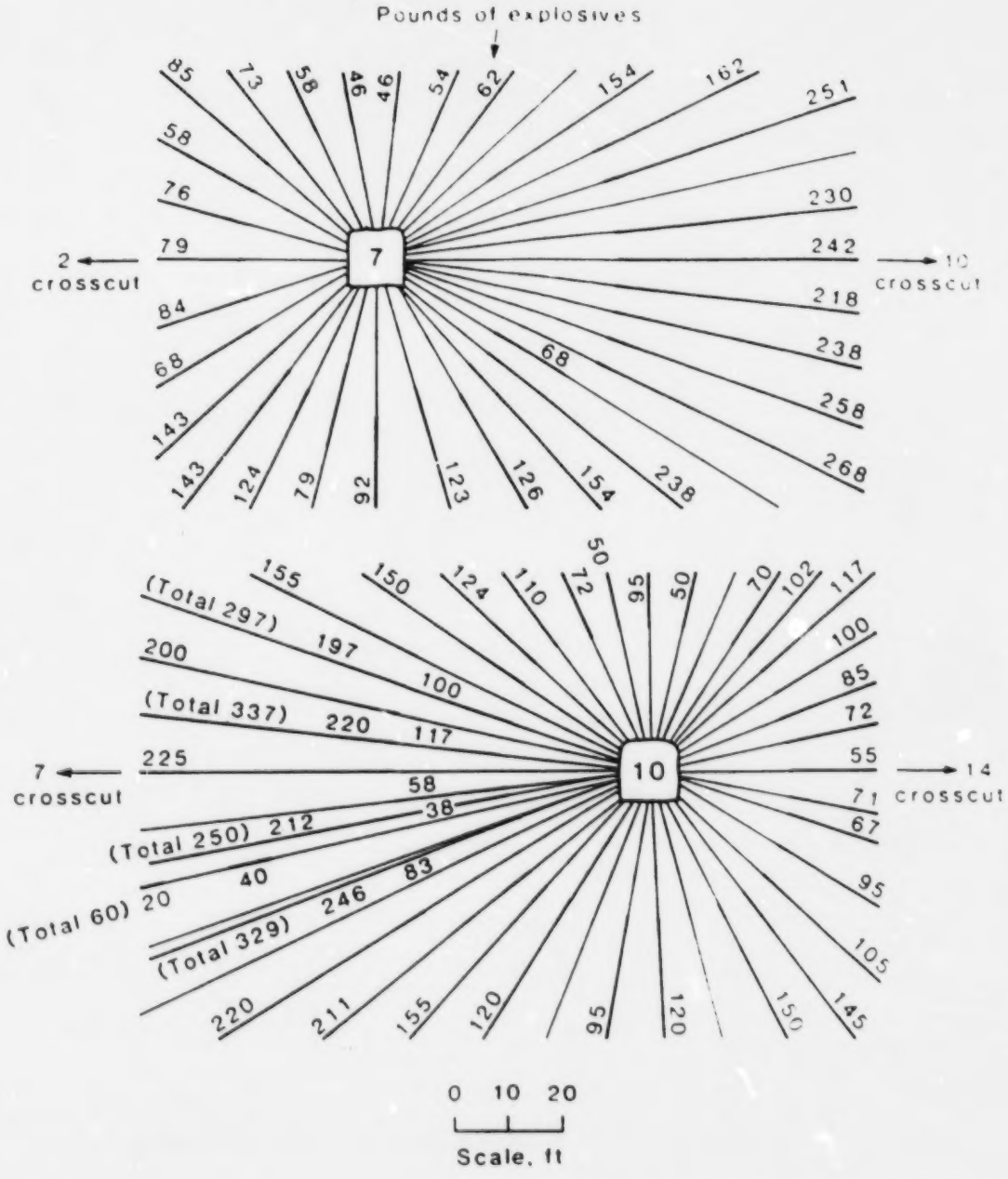

FIGURE 5: Preconditioning drillhole and loading patterns for the 7700 level (after McMahon, 1988).

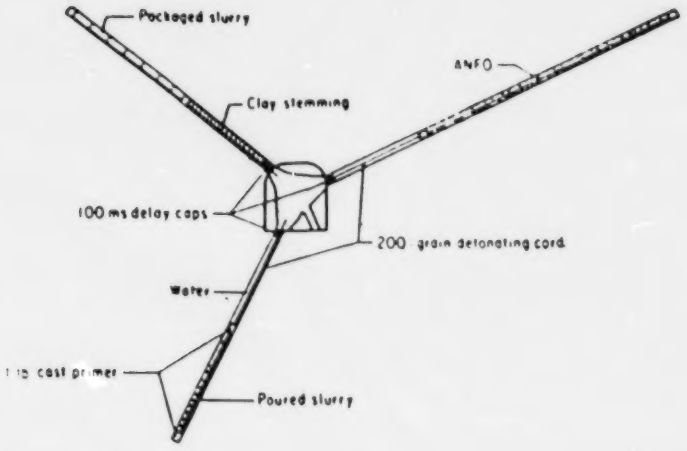

FIGURE 6: Typical loading holes (after Karwoski et al.).

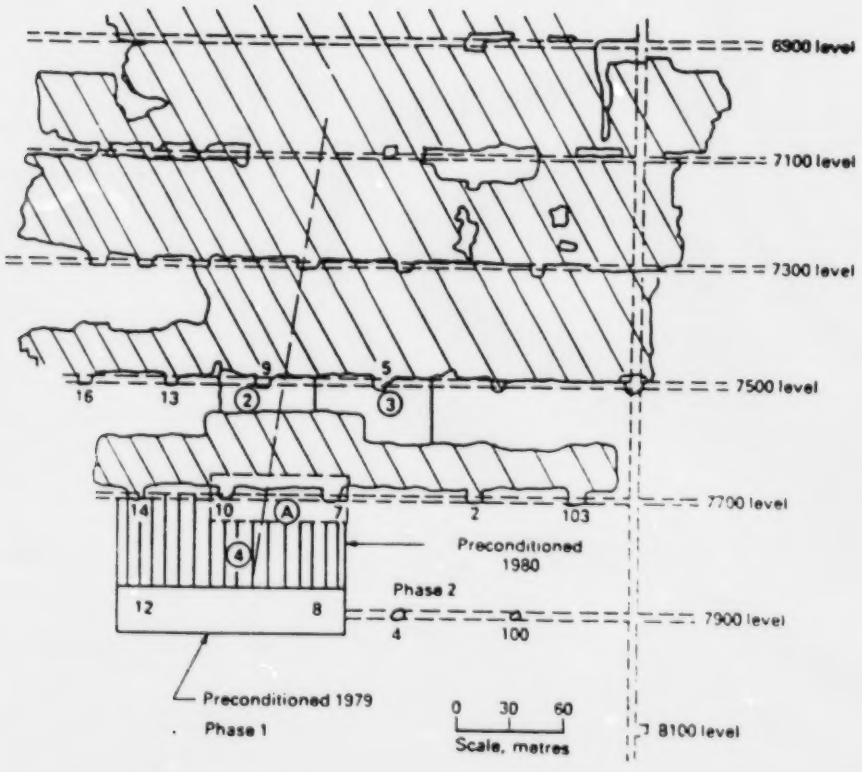

FIGURE 7: Section through a preconditioning testing zone of the Star Mine (after Blake, 1984).

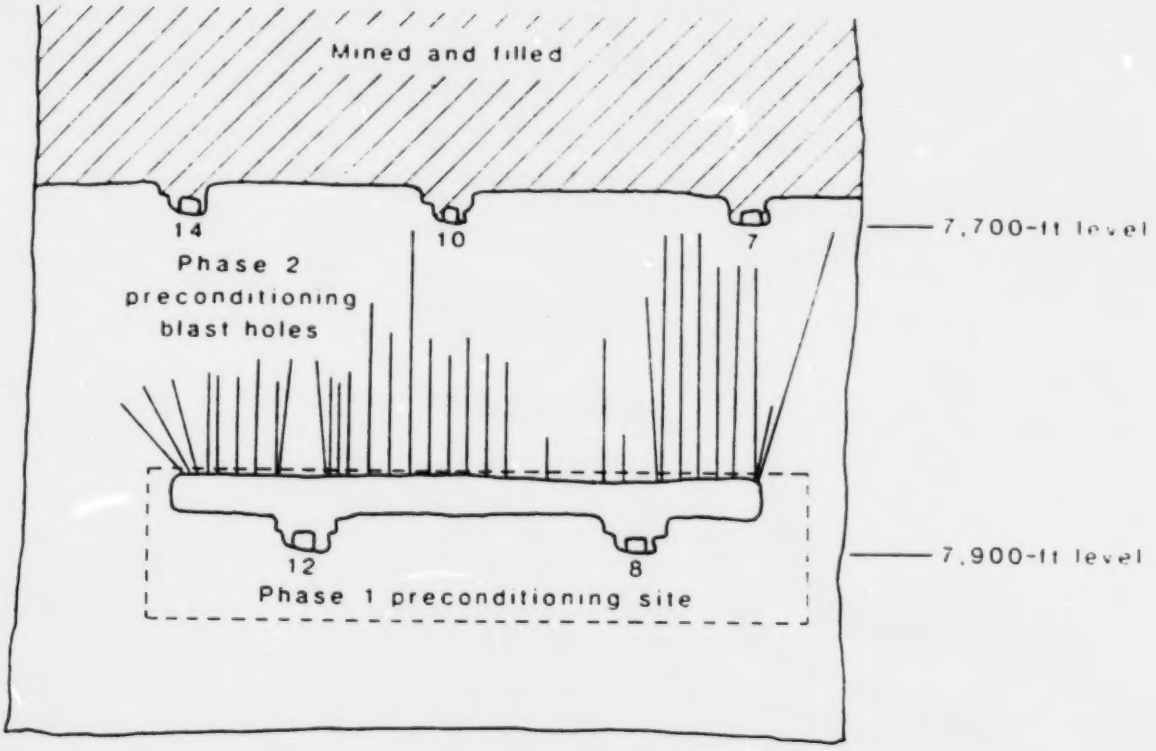

FIGURE 8: Preconditioning hole pattern of the 7900 level stope (after McMahon, 1988).

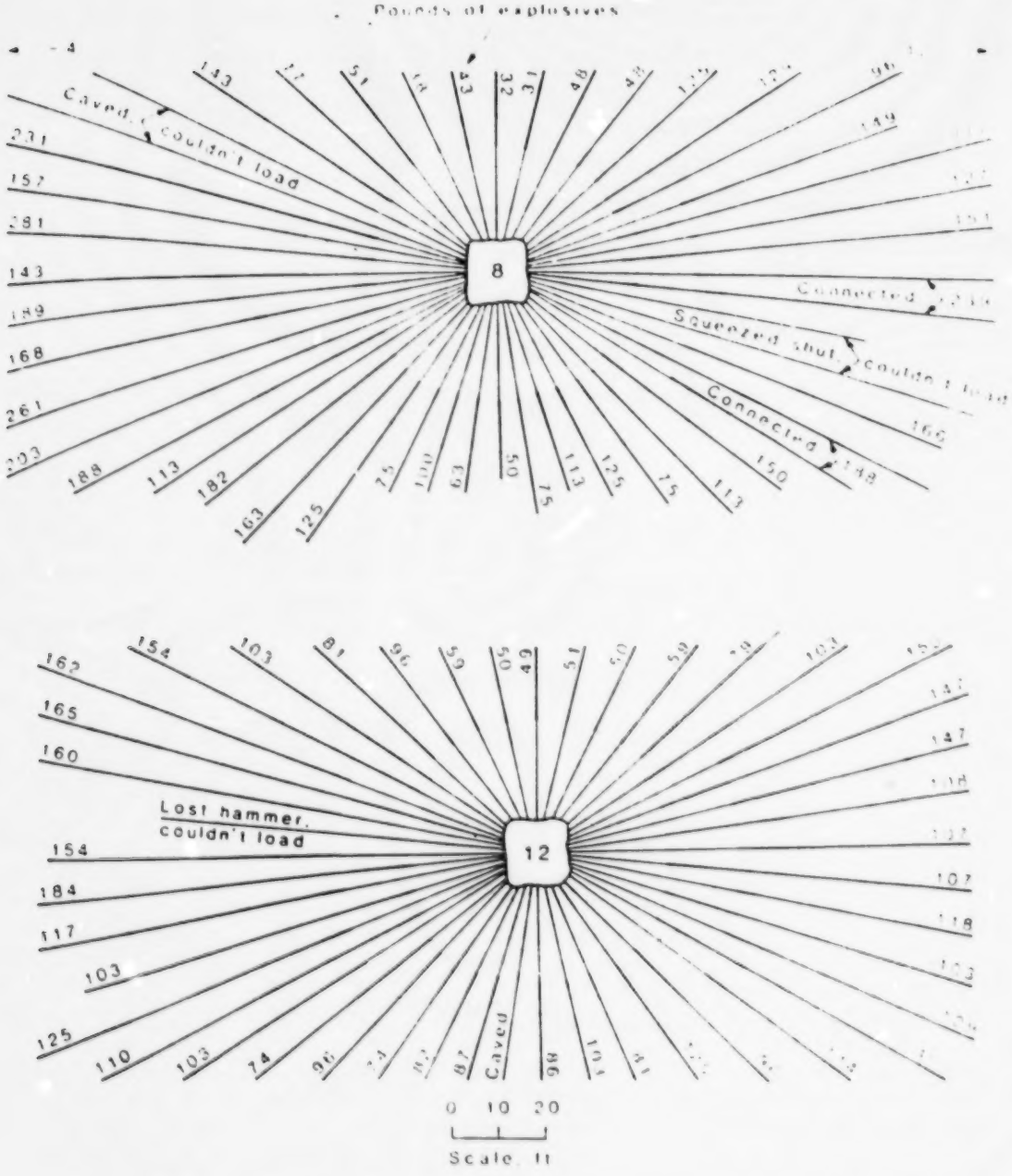

FIGURE 9: Preconditioning drillhole and loading patterns for the 7900 level (after McMahon, 1988).

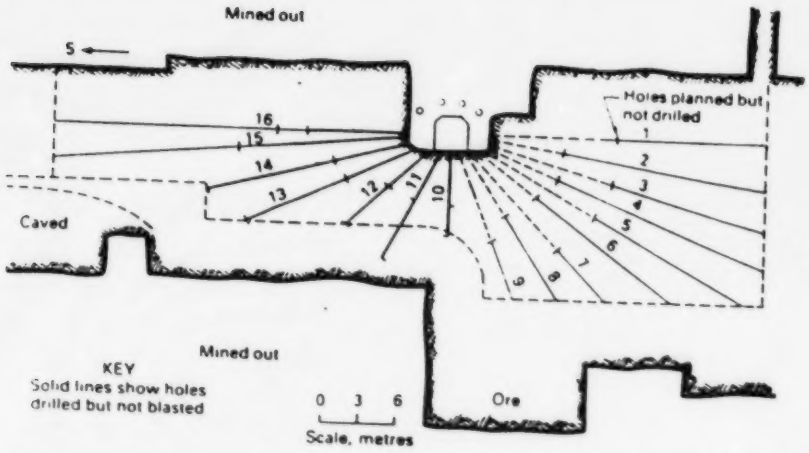

FIGURE 10: Preconditioning the 7500 level at the Star Mine (Blake 1984).

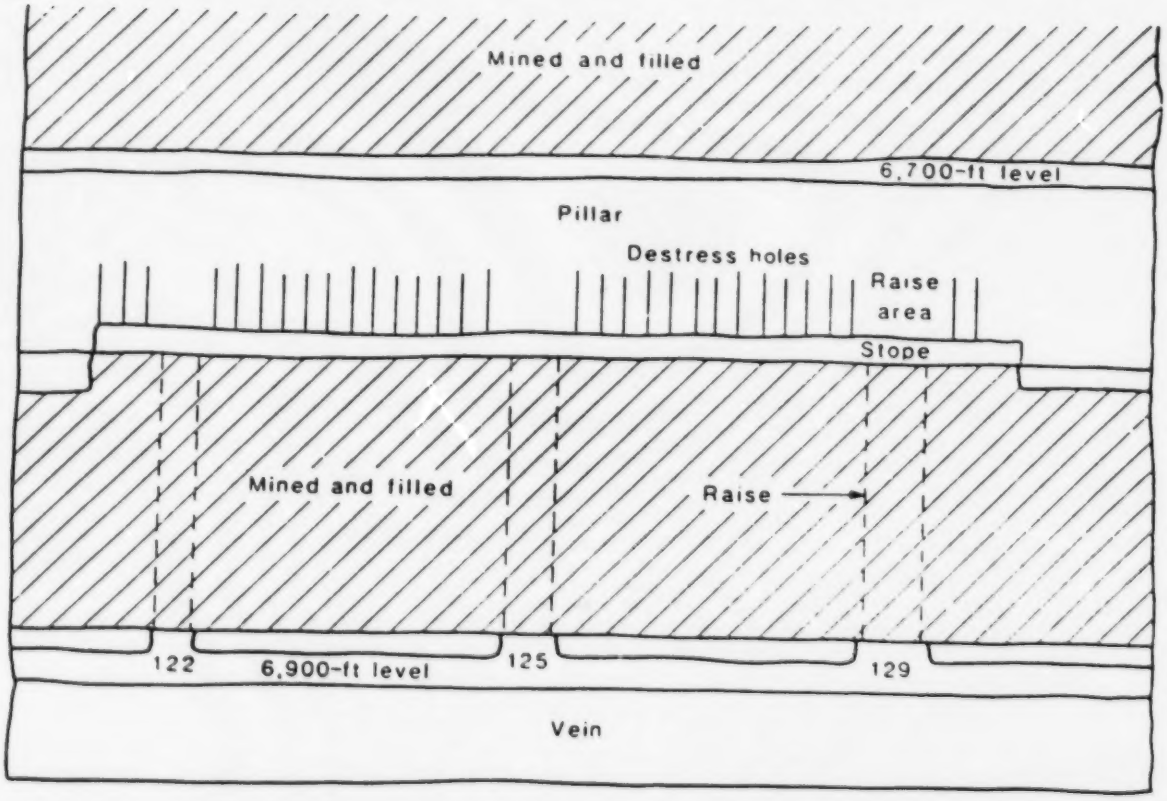

FIGURE 11: Preconditioning procedure used at the 6700 level of the Star Mine (after McMahon, 1988).